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E.L. 10/80 GREAT PYRAMID

AND

E.L. 12/78 SCAMANDER

SUMMARY REPORT

SEPTEMBER 1981

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D. A. CLARKE

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Where?
Note :- Plans attached to this report are
of a preliminary nature only.

NO PLANS EVER PRODUCED.

ALL INFORMATION PROVIDED IN OTHER REPORTS TO AGPT.
D.H.P.

Prof. R. Davis (consultant)

Distribution :- Exploration Dept., Camberwell (without plans)
R. Hine, Hobart (without plans)

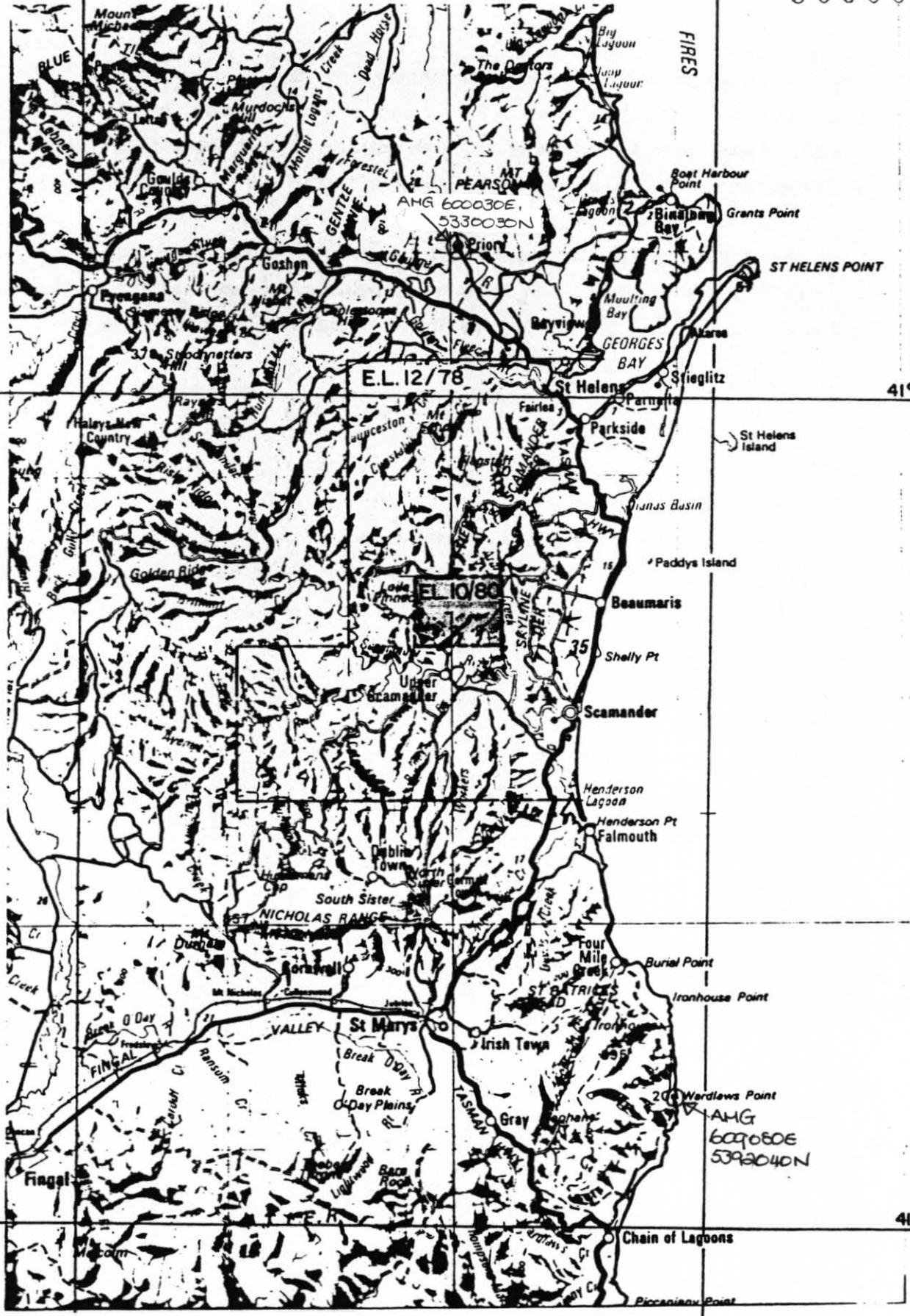
1. INTRODUCTION

Exploration Licence 12/78, Scamander, covers 276 km² in St. Helens - Scamander district of N.E. Tasmania (Fig. 1). It was granted to B.H.P. on 26th September 1978 and is renewable at 6-monthly intervals. A number of old mineral workings occur in the area, including the North Scamander and Great Pyramid prospects.

The Great Pyramid tin deposit was discovered in 1909 and has been tested intermittently by various individuals and companies. The prospect was unclaimed from 1975 to late 1976, when a 12 km² area was gazetted as exempt from the Mining Act. Special Reserve No. 236 was set up to enable the Mines Department to drill and assess the potential of the deposit. Following completion of this work the area was offered for tender in August 1979.

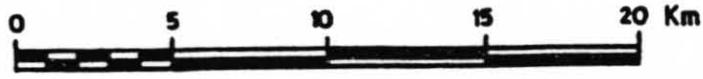
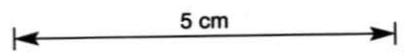
As we already held the surrounding area under E.L. 12/78 and considered Great Pyramid to be a very promising prospect, a proposed programme of intensive testing was submitted. This was aimed at delineating a large open-cut tonnage of low-grade tin mineralisation. Our tender indicated that we would spend at least \$400,000 on drilling in the first year and further work would be undertaken if warranted. We also offered to repay the Tasmania Mines Department the costs of their exploration.

Our tender was accepted and we were invited to apply for an Exploration Licence in February 1980. This was granted on 21st July 1980 for an initial period of 6 months. The licence is renewed every 6 months as long as the Mines Department are satisfied that active exploration is being carried out. There is no expenditure commitment in the licence document.



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Scale 1:250,000



Centre
Melbourne

Date
31-8-81

THE BROKEN HILL PROPRIETARY CO LTD.
E.L. 10/80, GREAT PYRAMID, TAS.
LOCATION MAP

Project No

Drawing No.
A4-2255

At present we are conducting on-going exploration in the Scamander licence area. Follow-up diamond drilling on the North Scamander prospect will commence at the beginning of October. At Great Pyramid our initial diamond-drilling programme was completed in June. Various difficulties were encountered and not as much was achieved as had been anticipated. We are presently conducting a major assessment of Great Pyramid and all data from previous work has been compiled. We now have a much better understanding of the three dimensional picture of Great Pyramid than has been possible previously.

Further exploration on this complex prospect will be costly. The size potential is apparently limited and the grade marginal. A decision will shortly have to be made as to whether we proceed with further testing of Great Pyramid, attempt to locate additional high-grade reserves from surrounding prospects, or farm-out the areas to another company.

2. REGIONAL GEOLOGY, STRUCTURE AND MINERALISATION

The sedimentary rocks in the area of interest consist of the Mathinna Beds, a sequence of sandstones, siltstones and shales showing turbidite structures. Massive beds of quartzite occur in areas of mineralisation - the silicification is probably an alteration effect. These sediments have been dated as Lower Devonian. The Mathinna Beds are intruded by a suite of granitic rocks of Upper Devonian age that represent the southernmost extension of the Blue Tier Batholith. A long, narrow vertical dyke of predominantly granodiorite porphyry extends southwards along Scamander Tier. Tertiary and Quaternary sediments are exposed near the coast.

There are two dominant structural trends. Broadly NW trending regional folding is evident with upright axial planes. Extensive faulting occurs on a similar trend. These fault zones are generally several metres wide, comprising many sub-parallel, steeply dipping fractures. The mineralised lodes that fill fractures in the Mathinna Beds generally trend NE but with a wide spread of orientation. They are broadly perpendicular to bedding and probably represent tensional fractures related to the regional folding. Figures 2 and 3 give some indication of the main lineaments.

The economic geology of NE Tasmania is dominated by the Blue Tier Batholith which lies north of the project area. Projections from the main granite massif are postulated to underlie the prospects of the Scamander area at depth. Most mineral occurrences consist of small, discontinuous hydrothermal quartz lodes of variable mineralogy in Mathinna Beds and granite rocks. A regional mineralogical zonation is evident with W and Sn mineralisation occurring in the west, Cu deposits in the central part of the area and Ag-Pb-Zn deposits in the east.

3. GREAT PYRAMID

3.1 Previous Work

Since its discovery in 1909 a number of individuals and companies have had an interest in mining leases at Great Pyramid. The prospect has been explored by adits, percussion drilling and diamond drilling. The main phases of mining and prospecting activity are summarised below :-

- 1909-11 Great Pyramid Tin Mines N.L. - Some 600 metres of adits driven into the E and W sides of the hill on two main levels, mostly parallel to the mineralised vein set. Channel sampling of adits; sampling of shafts and trenches.
- 1914 Troy Tin Syndicate - Resampled most of the adits but did no further work.
- 1925-36 Mr. H. Aulich - Only serious mining attempt. Total production 331t ore for 5.38t concentrate at an average recovery grade of 0.88%. Presumably most of this material came from the N. adit workings.
- 1945 Ore dressing investigations by Tasmanian Mines Department.
- 1957 At the request of the then owner Mines Department took 17 bulk samples from the workings. Assay results differed considerably from earlier sampling.
- 1963 Mines Department carried out further ore-dressing tests on 3 bulk samples from the N. adit.
- 1964-65 B.H.P. - 26 vertical percussion holes at 50 - 60m spacing. Only 3 holes intersected 'significant' mineralisation. (The definition

of 'significant' with respect to tin mineralisation has probably changed somewhat since 1965, with the low cut-off grades now being achieved in open-cut bulk mining operations.)

One 242m 45° diamond drill hole to test mineralisation in the N adit area - drilled oblique to the main veinlet set. Mineralisation intersected was sporadic. A ground magnetic survey was carried out.

1969-74 Paringa Mining and Exploration Co. / Aberfoyle Management - 137 vertical percussion holes to depths of up to 150 feet (45m) for a total of 5000m. Drilled on a grid pattern to test soil anomalies at a spacing of 50 feet (15m) on lines 100 feet (30m) apart. This drilling outlined two areas of mineralisation with grades \geq 0.30%. 6 vertical diamond-drill holes for 666m total in the C adit area SW of the summit. Best intersections were 41m @ 0.40% Sn and 33m @ 0.38% Sn.

1976-79 Tasmania Mines Department - 4 inclined diamond-drill holes, 60° depression, 710m total. Located on SW side of hill to test mineralisation in the C adit area. Holes drilled approximately parallel to the main veinlet set.

Of this earlier testing the Paringa - Aberfoyle vertical diamond-drill holes are considered to have been most successful in terms of returning reliable samples.

3.2 Rationale and Proposed Programme - 1980/81

The rationale behind our tender was based on the view that the potential viability of the Great Pyramid prospect has improved since 1965, due to changed economic conditions, improved mining methods and a better understanding of the geological controls of mineralisation. In addition much of the testing to date was considered to have been inconclusive. We believed the prospect had the potential to provide sufficient reserves for a major open-cut tin mine. A reserve of 4 Mt @ 0.31% Sn was assumed (Aberfoyle) and we were hoping to prove a 10 Mt resource.

Our testing programme was aimed at -

1. Accurately delineating such a deposit.
2. Establishing an accurate and reliable grade.

Our proposed work programme involved several stages, each dependent on the results of the earlier stage. It can be summarised as follows :-

Phase 1 - 2000m. HQ diamond drilling. 7 holes to investigate known mineralised blocks and possible extensions, 3-4 holes on major untested soil geochemical anomalies flanking the main areas of mineralisation.

Phase 2 - 4000m percussion or reversed circulation drilling to test the main mineralised blocks on a grid system.
Bulk metallurgical testing.

It was anticipated that Phases 1 and 2 could be completed in one year at an overall cost of \$500,000. Further work would be dependent on results.

Phase 3 - Bulk testing and additional drilling. This would include some deep drilling to investigate mineralisation at depth, closer to the presumed underlying source granites.

Phase 4 - Feasibility study.

It was stated in the tender that an important part of our proposed activities at Great Pyramid would be in integrating the work with our tin - base metals exploration programme on E.L. 12/78.

3.3 Summary of Exploration - July 1980 to date

Work carried out during the present phase of evaluation is summarised below :-

1. Aeromagnetic surveys of E.L.'s 12/78 and 10/80.
2. Re-logging of Aberfoyle and Mines Department drill core. Continuous plotting and assessment of earlier work.
3. Levelled grid established and base plans prepared at 1:1000 scale.
4. Geological mapping at 1:1000 scale on surface and underground, combined with a statistical analysis of vein sets. Detailed 1:200 scale mapping of adits. Regional mapping at 1:10,000 scale.
5. Soil geochemistry. Grid area sampled at 25m intervals and -40# soil fraction analysed for Sn, W, As, Cu, Pb, Zn and Ag.
6. Diamond drilling - 13 inclined holes completed to a total depth of 1156 metres.
7. Ground magnetics over grid area.

The first year's exploration costs were funded by the Resource Technology section. The Exploration Department has taken over responsibility for costs as from June 1st 1981.

3.4 Geology and Structure

Sandstones, shales and quartzites of the presumed Lower Devonian Mathinna Beds form the host to the tin mineralisation. The finer grained sediments are sheared, incompetent rocks which pinch and swell along strike. The sandstones and quartzites are thicker bedded, well jointed and fractured. Rock types are inter-gradational, due to variations in the proportions of each rock type, variations in thickness of bedding and degree of alteration. The quartzite is probably an alteration product of the sandstone. Mapped rock unit boundaries are thus somewhat subjective.

The Mathinna Beds are folded about approximately NW-SE axes and the evidence suggests that Great Pyramid lies on the upright limb of an anticline, with its hinge to the NE. Detailed mapping, both on surface and in adits, has defined two structural domains separated by a NW striking fault. Drilling indicates that this fault has a steep dip to the NE. South of the fault bedding strikes approximately NNW and dips at 60-70° to the SW, north of the fault dips are more variable and locally flatter. No positive correlation can be made across the fault although the two main quartzite units may be an extension of one another.

Geometrically the joints are normal to the fold axes, strongly implying that they are related to folding. Measurements by Aberfoyle in boreholes GPY 1, 3, 5 and 6 on the SW side of the hill show the average dip of the fractures to be 60°.

A weathered diorite dyke some 5 metres wide intrudes the Mathinna Beds at Great Pyramid. It trends ENE, sub-parallel to the dominant (mineralised) fracture direction, and post-dates both fracturing and mineralisation. This dyke represents the only known igneous activity in the immediate mine area.

3.5 Mineralisation

Mineralisation is in the form of cassiterite which occurs as disseminated grains and aggregates in an ENE trending quartz-veinlet system. The host rock Mathinna Beds are basically unmineralised. The joints are planar and generally dip at 60-70° to the NW; they range up to 5mm in width and exhibit open space filling. The general sequence of crystallisation within the veins is muscovite → cassiterite → quartz → sulphides. Minor wolframite, fluorite and pyrolusite have been recorded.

Fracture density ranges up to 120 per metre in the more intensely mineralised areas. The mineralised fractures exhibit siliceous alteration haloes usually 1 - 2 cm wide. When the density is such that these haloes overlap the quartzite rock type is formed. The more competent sandstone beds probably developed more continuous discrete fractures than the shale during folding, thus allowing the easier passage of mineralising solutions. Where a vuggy quartz-cassiterite-sulphide fissure in the quartzite enters a shale bed it often persists only as a limonite-stained fracture line.

Drilling to date has defined two main zones of mineralisation, an East and a West block. These are interpreted as being truncated by the NW trending fault and it is possible that they are faulted repetitions of the same quartzite unit. The apparent difference in strike of the two mineralised

lenses is a product of the steep topography on the SW side of the ridge; both lenses are parallel to bedding. The most important areas of mineralisation appear to be in close proximity to the interpreted fault and drilling has indicated that the fault itself is mineralised. Some intersections of massive sulphide mineralisation carrying tin values up to 2.8% have been encountered in drilling (MD3, DDS1, BPD7).

To summarise the controls on the parallel-veinlet style cassiterite mineralisation at Great Pyramid are as follows :-

1. The presence of thick sandstone/quartzite units. Bedding generally has a variable SW dip.
2. Dense, closely spaced fracturing. Fractures are oriented normal to the local fold axes; they trend ENE and dip fairly steeply to the NW.
3. Proximity to a through-going NW trending fault.

An environment of mineralisation has thus been formed by the above controls, in combination with a regional zoning effect related to a buried granite source for the tin.

3.6 Géochimie

Following acquisition of the Exploration Licence a survey grid was established and the main area of interest soil sampled at 25m intervals on lines 50m apart. This work was designed to give indications of extensions to known mineralisation. The main anomalies are outlined in Figure 5 and it can be seen that the West Block is particularly well defined. Lower order anomalies on the north west side of the East Block have been tested during the recent drilling programme. A high-order tin anomaly

(+ 5000 ppm) west of Brock's shaft has not been adequately tested.

Transport of anomalies occurs over the steep terrain and this is particularly evident for the more mobile elements. Comparison of anomaly plans for the various elements also shows evidence of mineral zoning. Drilling has located areas of relatively high silver values, peripheral to the main tin mineralisation.

3.7 Geophysics

A ground magnetic survey was carried out by B.H.P. during its 1964 programme. Magnetic axes can be correlated with pronounced jointing in the Mathinna Beds. Anomalies were explained as being due to the presence of minor amounts of iron-oxides and sulphides which are present in the quartz-cassiterite veinlets. One linear anomaly was clearly associated with the ENE striking diorite dyke which crosses the hill. Other NW trending lineations on the anomaly plan may be related to faulting.

In view of the interest in locating fault-controlled massive sulphide mineralisation, and because of difficulties in tying in the earlier work to the present metric grid, a new programme of ground magnetics has recently been completed. Traverses were run NE-SW to locate any fault and/or bedding controlled mineralisation, and NW-SE to locate any significant disseminated mineralisation in the veinlet systems and to assist mapping of the diorite dyke. Results are currently being compiled.

An aeromagnetic survey of the Scamander licence area shows up a minor anomaly just north of Great Pyramid. This cannot be fully explained, though there is some indication of a linear grouping of weak anomalies along a NE trend which may be due to a basic dyke.

3.8 Diamond Drilling - 1980/81

The drilling programme as originally planned was to consist of 2000 metres of angled diamond drilling and 4000 metres of percussion drilling. Previous investigations were considered to have been compromised to some extent by drilling sub-parallel to the trend of the vein system. The optimum drill hole orientation would seem to be at the highest practical angle to both bedding and the mineralised jointing. Boreholes were sited to drill at an optimum direction of 120° M, inclined at 60° on the southern side of the fault (where bedding tends to be steeper) and 70° on the north side. All core was logged, photographed, split using a diamond saw and sampled. It is presently stored on site.

The programme was planned to centre on the two areas of established mineralisation, to check grades and investigate lateral and vertical extensions. In the event drilling problems and consequent failure to reach anticipated depths meant that only two boreholes tested the known higher-grade areas. Some 13 holes were drilled between 17th October 1980 and 25th June 1981. Of these only 5 were successfully completed. A total of 1228.69 metres of drilling was achieved, including 148.19m of percussion drilling. The main problems were caused by the open-space fracturing. Complete water loss was commonly experienced and similar problems occurred during percussion drilling. Paringa-Aberfoyle reported early termination of a number of their holes due to loss of compressed air due to fractures. These technical difficulties reflected themselves in an unacceptably high cost (approaching \$200 per metre in our recent programme). The extremely slow rate of progress also greatly increased overall costs, particularly in regard to salaries for support staff.

The planned grid percussion drilling programme to more accurately outline and obtain grades of reserves in the

two mineralised blocks was not carried out. This was partly because results from the diamond drilling did not upgrade the prospect, but mainly because the costs and timing were greatly under-estimated.

Because of the difficulties in reaching planned depths, and the cutting short of the proposed programme, this latest phase of diamond drilling has accomplished relatively little in upgrading the deposit or altering previous reserve estimates. No new zones of + 0.20% Sn were delineated, although numerous intersections in the range 0.10 - 0.20% were obtained. However the drilling has significantly increased our overall knowledge of the deposit in terms of detailed geology, structure and controls on the mineralisation.

In view of the problems encountered in drill testing the Great Pyramid deposit to date the planning and costing of any future drilling programme will need careful thought. In addition the highest standards of technical competence will be required from drilling contractors.

3.9 Metallurgical Recovery

A preliminary assessment of Great Pyramid by our Resource Technology section has indicated that, given certain minimum reserves and grade, the project is particularly sensitive to metallurgical recovery. Milling and concentrating will be the largest sector in both capital and operating costs. The viability of the project would thus be directly related to the mill recovery and the grade of concentrate produced.

A detailed metallurgical programme was proposed to test Great Pyramid ore at B.H.P. Whyalla. Four samples of oxide ore at various grades (0.30 - 0.65% Sn) and exceeding 200 kg in weight were submitted for testing in September 1980. Various modifications have since been made to the testing programme. Because of staff turnover and other

priority work results are not yet to hand. An additional bulk sample of 400 - 500 kg has recently been collected and submitted to the Mines Department in Launceston for similar test work.

Several earlier ore-dressing investigations have been carried out but more definitive results are still required. In 1945 the Tasmania Mines Department did some work on a sample of ore assaying 0.64% Sn. It was concluded that most of the cassiterite is readily recoverable even at a -36# grid. A recovery of 78.6% was reported. Further work was done in 1963 on three bulk samples with grades of between 0.47 - 2.17% Sn. This investigation was basically carried out to determine the recovery of tin attainable by tabling. Ores were initially ground to -40# and coarse tailings from the tabling were later reground to determine the additional recovery of tin possible with finer grinding. Recoveries from the initial grind ranged from 60% for the lower-grade ore to 80% for the high-grade ore.

In 1971 a composite sample from percussion drilling in the oxidised section of the orebody was tested, again at the Mines Department. There are some doubts about reliability - the grain size of cassiterite may have been unnaturally degraded during the drilling process. The sample assayed 0.29% Sn and a concentrate of 47.6% Sn was obtained with a recovery of 63.5%. It was concluded that a recovery of 60 - 70% could be expected in commercial treatment of this type of ore.

Ideally the cassiterite, upon fine crushing or coarse grinding, should be amenable to liberation at fairly coarse particle size such that beneficiation by gravity methods is possible. A very high recovery should be technically feasible by using modern, sophisticated gravity concentration methods and cassiterite flotation. However there is always the problem of balancing extra capital and operating costs

against the increased revenue from higher recovery. To increase overall profitability it is often necessary to settle for lower recovery and a shorter mine life.

The Resource Technology group have assumed a 65% recovery in their preliminary feasibility calculations. Our general feeling is that a satisfactory recovery should be obtained for the oxide ore and that this figure may be a little low. In the longer term, however, it is probable that to obtain a resource of suitable size it will be necessary to locate and prove higher grade sulphide ore to supplement presently known oxide reserves. This ore will be more difficult to treat and will again necessitate lengthy metallurgical test work.

It could be said that mineral dressing and metallurgical considerations are only partially relevant at this stage of the investigation. This is if we accept that Great Pyramid ore is not likely to be inherently more difficult to treat than ore of similar type tested elsewhere. The main factors at present would seem to be tonnage and grade, and the possibilities of increasing our present figures.

3.10 Reserves and Assessment

Prior to our recently concluded drilling programme it was considered that the Great Pyramid prospect represented a significant open-cut resource with potential reserves in excess of 4 Mt of low-grade, easily recoverable cassiterite mineralisation. Our hope was to increase reserves to some 10 Mt at around 0.30% Sn. Economies of scale would also have made a larger reserve (15 - 20 Mt) at lower grade attractive.

A problem with assessing earlier work on the prospect is the fact that most adits and several diamond-drill holes

were driven parallel to the mineralised veinlet system, producing obvious difficulties for sampling and grade estimation. Previous testing was therefore considered to have been inconclusive. This was not due entirely to inefficiency or bad exploration. The prospect is inherently difficult to prove. There are several different controls on mineralisation and open fractures in the main mineralised zones cause technical difficulties. As already seen our recent drilling has brought home some of the problems associated with investigating this deposit.

Not as much information was obtained from the diamond drilling programme as had been expected. However, the objective of attaining 10 Mt at 0.30% Sn has been negated. Recent drilling has not increased previously indicated 'higher-grade' areas and has not located new areas. Possibilities for additional reserves at depth still exist but a preliminary impression is that overall grades tend to diminish with depth.

In our updated reserve calculations we have used all available data from work to date. Level plans have been drawn up at 10 metre intervals. Grades are assigned to each intersection on the relevant level by weighting values over the 5 metre interval above and below the level. A polygonal area of influence method has then been used to obtain areas and hence volumes assigned to each intersection - this is necessarily somewhat subjective. There are various alternative methods of calculating reserves and it is accepted that the method used is only a preliminary one. However, it is not anticipated that more sophisticated methods would give significantly different results based on the data available.

Reserves have been classified as either Probable, Possible or Potential (instead of the normal proven/probable/possible or measured/indicated/inferred). This indicates that there is a lower level of confidence in the reserves

than would normally be the case. Because of the complexity of Great Pyramid a large number of additional boreholes would be required before reserves could be said to be proven.

A preliminary reserve calculation, based on the data from 137 percussion holes, was carried out in 1971 by Aberfoyle. This is summarised in the table below, in combination with our latest calculations.

TABLE 1

GREAT PYRAMID PROSPECT - SUMMARY OF RESERVES

(210 Level to 90 Level)

Million tonnes / Grade (Sn%)

	<u>Probable</u> (indicated)	<u>Possible</u> (inferred)	<u>Total</u>
<u>WEST BLOCK</u>			
B.H.P.	2.22/0.22	0.43/0.25	2.65/0.22
Aberfoyle	1.00/-	1.97/-	2.97/0.33 *
<u>EAST BLOCK</u>			
B.H.P.	1.01/0.21	0.44/0.20	1.45/0.21
Aberfoyle	0.40/-	0.70/-	1.10/0.28 *
<u>TOTALS</u>			
		B.H.P.	4.10/0.22
		Aberfoyle	4.07/0.31 *

It can be seen that Aberfoyle were more optimistic with regard to grades. Our recent calculations are more optimistic as regards the ratio of probable to possible ore but the overall reserve figure is remarkably consistent. It should be stated that in our preliminary calculations there was no attempt made to define 'mineable' reserves - the figure of 4.1 Mt above is therefore a maximum figure at this point in time.

Resources have indicated that a reserve of less than 5 Mt is considered to be too small for B.H.P.

Further work would therefore be partly focussed on increasing reserves. However, grade is probably a more critical factor in that a larger percentage increase is required to approach the 0.30% target. Our method of investigating reserves, by effectively slicing the hill apart at 10 metre intervals, immediately indicates areas where information is lacking and where potential exists for increasing reserves and grades.

Any further testing of the deposit must attempt to reduce the unknown factors to the point where a stable, long-term mining operation can be planned. This is going to be extremely difficult. The various different controls on mineralisation at Great Pyramid tend to limit the higher-grade ore to 'blocks' of limited volume. Blocks are limited laterally by the width of the most intensely fractured zones, and vertically by boundaries between the quartzite units and shale or interbedded shale and sandstone. Faulting in more than one direction causes additional structural complications. At Pyramid any reserve calculation and assessment is also rendered difficult by erratic grade variations over short distances; this is obvious both in channel sampling and in redrilled boreholes, where values reflect only broadly those encountered in the original holes. The deposit can therefore never be proven to the same confidence level that a more predictable orebody can.

An element of doubt will always be present in reserve calculations and to minimise this as far as possible a considerable amount of drilling would be required.

In addition to proving additional oxide reserves in the main part of the deposit two other possibilities occur. Soil sampling has shown a relatively untested area to the west of Brock's Adit, where values range up to 0.66% Sn and compare favourably with results from the two main blocks. Aberfoyle evidently considered the area to contain only small scattered pockets of tin but some further investigation might be warranted. Another possibility is provided by promising massive sulphide intercepts in boreholes MD3, DDS1 and BPD7 (Fig. 4). These appear to coincide with an elongate ground magnetic ridge running broadly along the NW/SE trending fault zone. A continuous mineralised fault zone, several metres in width, would certainly serve to increase reserves and the overall potential of the deposit.

Reasonable silver values were obtained from some parts of the deposit during recent drilling. Unfortunately the highest silver values occur in areas marginal to the best tin mineralisation - this is probably a mineral zoning effect. No other recoverable metals occur in the oxide ore.

4. NORTH SCAMANDER

North Scamander is one of a number of small prospects in E.L. 12/78. Surface mineralisation is primarily Ag-Zn, with minor Pb and Cu. The mineralised zone trends NW-SE and mineralisation appears to be partly replacement of shale horizons and partly fracture fillings.

An earlier investigation was carried out by The Electrolytic Zinc Company of Australia Ltd. in the late fifties. Magnetic and electromagnetic geophysical surveys were carried out and one diamond drill hole intersected 1.83 m @ 4.44% Zn and 34 g/t Ag. It was concluded that the mineralisation was too sparse and low grade to warrant further testing.

4.1 Summary of Exploration - 1978 to date

A helicopter-borne aeromagnetic survey was flown over the Scamander licence area in 1979. This disclosed a strong magnetic anomaly coinciding with the North Scamander deposit. Magnetite and pyrrhotite are abundant in the deposit and in this respect it is apparently different to other prospects in the area. Anomalous tin values were obtained from surface soil and rock chip samples. Regional mineral zoning theory indicates a probable increase in tin grade with depth, and the abundance of pyrrhotite bears a possible comparison with some of the sulphide tin deposits of NW Tasmania. It was therefore decided to investigate the prospect in some detail.

In addition to the aeromagnetics a Dighem survey of part of the licence area was flown, but this failed to reveal any major, near-surface massive sulphide targets.

Work at North Scamander to date has included gridding and soil sampling, accompanied by geological mapping. Four shallow percussion holes tested soil geochemical anomalies

with poor results. Geophysical work has included detailed ground magnetics, downhole borehole logging (Sirotem) and magnetic susceptibility measurements on core. Results from the geophysics are currently being assembled into a comprehensive report. A target zone with a strike length of around 400 metres has been defined with a steeply dipping anomaly.

4.2 Diamond Drilling

To date three diamond drill holes totalling 610 metres have tested the North Scamander geophysical/geochemical anomaly. A 5 - 15 metre wide, steeply dipping breccia zone containing massive matrix pyrrhotite-magnetite-chlorite mineralisation was intersected. Best values obtained were from borehole NS2, i.e. 1.2m @ 0.93% Sn, 28 g/t Ag (within 9.4m @ 0.22% Sn) and 2.0m @ 0.46% Sn, 64 g/t Ag (within 9.2m @ 0.20% Sn). Minor base metal stringer mineralisation was observed throughout, suggesting that a broad fault zone is present.

Following completion of the initial drilling a detailed ground magnetic survey and downhole logging programme was carried out. This has indicated that of the three holes drilled to date only NS2 tested the full width of the geophysically anomalous zone. NS1 tested the zone in part only and is to be deepened in a renewed drilling programme which will commence shortly. NS4 is to test the western part of the anomaly, drilling in the opposite direction to NS3 (which failed to intersect the breccia zone). Once the attitude of the mineralised breccia zone is clearly defined deeper drilling will be necessary to test possible higher-grade mineralisation at depth.

5. OTHER PROSPECTS

Fig. 2 shows some of the main prospects in the Scamander licence area and gives an idea of the broad regional mineral zoning. Broadly NW and NE trending lineaments dominate the geological map and are also evident on the magnetics (Fig. 3).

The Pinnacles prospect occurs to the NW of Great Pyramid and the mineralisation is similar, with fine-grained cassiterite in fractures in the quartzite beds. Six percussion holes were drilled by B.H.P. in 1965 but no significant mineralisation was encountered.

Most mineral occurrences in the region consist of small, discontinuous, hydrothermal quartz lodes of variable lithology. As such they are not really promising unless there is a coincident magnetic (or EM) anomaly. The Cramps target area consists of a NNE trending line of gossan cappings. It has a broadly coincident aeromagnetic anomaly trending NNW which is apparently offset by a major NE trending structure. A grid has been established and mapping, soil sampling and ground geophysics are planned. Preliminary estimates suggest a source at around 200 metres for the anomaly.

To the east of North Scamander Paul Beahr's prospect also broadly coincides with a magnetic anomaly. No work has been done on this prospect to date.

6. DISCUSSION

6.1 General

At present we are in the situation where we have calculated reserves at Great Pyramid (probable plus possible) of 4.1 Mt @ 0.22% Sn. Resource Technology indicate that our minimum reserve should be 5 Mt and for this minimum a grade of approaching 0.30% Sn is required. Metallurgical test work currently in progress is not going to reduce these target figures appreciably as far as a B.H.P. operation is concerned. Given suitable recovery results we have various alternatives open to us. These can be summarised as follows :-

1. Attempt to upgrade reserves at Great Pyramid, both as regards tonnage and grade, by further drilling (+ aditing ?). Change 'probable + possible' status of reserves to 'proven + probable'.
2. Accept that we are unlikely to prove a sufficient resource based on Great Pyramid alone. Search for additional ore elsewhere in the Scamander licence area. This could be added to the higher-grade near-surface reserve at Pyramid to provide a sufficient overall tonnage to satisfy a central plant.
3. Farm-out Great Pyramid. Continue with our on-going exploration programme in the Scamander licence area.
4. Farm-out both Great Pyramid and Scamander.

6.2 Great Pyramid

As far as Great Pyramid itself is concerned the following factors are considered relevant :-

6.2.1 Plus factors

1. We now have a good overall knowledge of the deposit, understand the main controls on mineralisation, and can see where possibilities for further reserves exist. It should be relatively easy to prove, say, 2-3 Mt of suitable reserves.
2. Some increase to the calculated grade could be anticipated on mining. Sludge assays during the recent drilling programme were usually higher than the core, indicating that some cassiterite is being lost during the drilling. The statistically erratic distribution of higher-grade patches would also tend to increase the final grade.
3. The overall grade can also probably be increased to some extent by the inclusion of higher-grade sulphide mineralisation associated with fault zones.
4. We have the infrastructure and expertise immediately available for further detailed exploration work.
5. Present reserve and grade estimates are probably sufficiently promising to attract a company interested in a smaller-scale mining operation than that envisaged by B.H.P.

6.2.2 Negative factors

1. The tonnage is clearly limited. We will have to work hard to prove 5 Mt @ 0.30% Sn. Is further work and considerable additional expenditure warranted for this size of deposit?
2. The recent drilling programme did not increase the + 0.20% Sn reserves.
3. Drilling to date has been technically difficult, slow and costly. Vertical drilling might be easier for

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future work but is not satisfactory where the dip of fractures exceeds 70° .

4. The geological and structural complexity of Great Pyramid, combined with significant grade variability, make it a very difficult prospect to satisfactorily prove.
5. A very large amount of additional drilling will be required to locate further reserves and prove the deposit to the required confidence level. The fact that to date 190 boreholes have only indicated a probable reserve of 3.2 Mt @ 0.22% Sn gives some idea of the additional drilling required.
6. The presently calculated grade is well short of that required. Indications are that the two main higher-grade blocks lack vertical continuity. Few 'high-grade' intersections have been made on the lower levels.
7. There are no other recoverable metals to sweeten the deposit. (Ag is locally high but the best Ag does not correspond with the best Sn.)
8. There would be major differences between the oxide ore and sulphide mineralisation as regards treatment. Sulphide ore would be more difficult to treat, with lower recovery.
9. Staff requirements - Pyramid is labour intensive because of its complexities. Could these geologists be put to more productive use elsewhere?

6.2.3 Possible further drilling

Additional drilling on Great Pyramid would not be done in one phase. There would be several steps to allow for continuous decisions as to potential at different stages of the testing:-

1. Drill prove higher-grade blocks. Any mining operation would commence on the near surface best mineralised areas to obtain a quick return on investment. Additional reserves at depth are probably irrelevant if this surface reserve is not accurately delineated and a reliable grade obtained. A statistical study is required to indicate the amount of drilling necessary.
2. Increase reserves -
 - (a) Test depth extensions of known higher-grade blocks.
 - (b) Test new areas - soil geochemical anomalies and blank areas of knowledge evident from studying the level plans.
 - (c) Test for sulphide mineralisation by drilling perpendicular to faults.

Further metallurgical testing of bulk samples will be required at some stage, particularly if sulphide ore is going to be a significant factor.

6.2.4 Costing

It is important to be honest about costs and fully realise the implications involved in going further with detailed testing of this deposit. Previous forecasts of expenditure on Great Pyramid have been greatly underestimated. The suggested costing for a two-stage assessment is as follows :-

1. Estimated expenditure to prove the two higher-grade blocks and obtain a good indication of the extent of additional reserves :-

	A \$
Percussion drilling (? vertical)	
E. Block 35 x 60m holes @ \$30/m	63,000
W. Block 30 x 60m holes @ \$30/m	54,000
Analyses	33,000
Diamond drilling for additional oxide reserves	
1500m @ \$150/m	225,000
Analyses	18,000
Diamond drilling for potential sulphide reserves	
1000m @ \$150/m	150,000
Analyses	12,000
Labour/overheads - say \$8000/month for 15 months (2 full time geologists + support staff)	120,000
Miscellaneous - computerisation of data, some metallurgical work etc.	75,000
TOTAL	750,000

2. Further expenditure if deposit is still promising

Diamond drilling to prove additional oxide and sulphide ore reserves, analyses, labour and overheads	750,000
Further bulk sampling, metallurgical testing, feasibility study	1,000,000

The above costs are pure guesswork but it would seem that a figure of around \$2.5 M will be required before we would have all the necessary information. As an order of magnitude figure this is probably not too far out - Newmont have spent about \$4 M on their parallel-veinlet style Taronga deposit and are still at the feasibility stage. Admittedly Taronga is a larger deposit (nearly 20 Mt) and there was less previous information. However, this is partly balanced by the mineralisation being more consistent and drill provable with good strike and dip continuity.

The basic question is whether it is worth getting involved to the extent outlined above when the potential reward is only a medium sized orebody of marginal grade.

6.3 Regional Approach

If we accept that Great Pyramid is unlikely to provide a sufficient resource on its own, then the emphasis changes to a regional evaluation. It is probably too early to consider farming out Great Pyramid until more is known about the regional potential.

In addition to North Scamander, where drilling will shortly recommence, other interesting prospects have been located. There appears to be a good chance of locating reserves of higher-grade sulphide mineralisation which could be combined with reserves at Great Pyramid to give an overall resource of suitable grade and tonnage to satisfy a central plant.

Focussing on these outlying prospects provides the opportunity for important new discoveries. A limited number of boreholes would give us a considerable amount of new information as to the potential of the area as a whole.

North Scamander is at a reasonably advanced stage but deeper drilling is required to test regional mineral zoning theories which indicate an increase in tin grade with depth in the sulphide-rich breccia zone. The Cramps grid area is promising with a coincident aeromagnetic anomaly and good values from sampling of surface gossans. Other aeromagnetic anomalies require follow up.

Costs for this phase of exploration would be reasonable. Preliminary gridding, ground geophysics, geological mapping and some geochemistry would be necessary. This would allow for the best possible siting of planned boreholes. Some 1500 - 2000 metres of NQ diamond drilling would allow us to test several targets and also drill a few deep boreholes. A further assessment would follow completion of this drilling programme.

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GREAT PYRAMID PROSPECT:
MEMORANDUM REPORT ON
ALTERNATIVES FOR SECOND
STAGE TESTING

R. Hine
Hobart

October 1981

1. INTRODUCTION

The principal aim of future testing at Pyramid should be to establish whether or not the grade as determined from previous drilling is a realistic one. This report covers a range of possible alternatives for second stage testing of the Pyramid prospect.

2. ALTERNATIVES

1. Pattern drill north and south blocks on 25m centres to the 100 level with vertical percussion holes. This would involve the drilling of approximately 95 holes for a total of 8000 metres. Direct drilling costs of the programme would vary between \$250,000 (\$30/m) and \$400,000 (\$50/m).

Advantages: - a 6 to 8m T block of ground would be tested;

Disadvantages: - reliability of the grades would still be in question;

- drilling is likely to encounter the same problems of previous programmes if the holes are not cased;

- will not reveal much in the way of structural ~~and~~ or lithological data;

- will not test any ground outside presently defined main blocks.

2. Drill out a small block of ground (~2000 tonnes?) at 2-5m spacing then mine and mill the block in an attempt to reconcile grades determined from previous drilling with ~~actual values~~ *head grade values*.

Advantages: - should enable an empirical correction factor to be applied to grades obtained from previous drilling;

Disadvantages: - result may not be applicable to the entire deposit.

3. Bulk sample existing underground workings and develop new drives at right angles to the vein sets - put material obtained through a treatment plant as above. Approximately 850-900m of drives from the existing adits would be necessary to adequately test the north and south blocks. The cost of driving the adits would be in the order of \$300,000 at \$350/m.

Advantages: - will give reliable grade estimates to compare with grades from previous drilling;

- will increase underground exposure and improve knowledge of the deposit geology;

- tin recovered could be sold to offset costs;

Disadvantages: - testing will be restricted to the 2 main levels through the deposit.

4. Target diamond holes to attempt to prove up small high grade bodies in previously untested areas (viz: geochemical anomaly SW of Brock's adit; mineralization along the fault; the magnetic anomaly north of the hill). This would involve approximately 1500m of diamond drilling for oxide reserves and 1000m for sulphide reserves. The programme would cost in the order of \$375,000 at \$150/m NQ coring or \$500,000 at \$200/m HQ coring.

Advantages: - would increase tonnage and grade potential of the prospect;

Disadvantages: - will not resolve problems relating to grades in the main blocks of mineralization.

5. Investigate the potential of either *a*) ore sorting by natural gamma radiation, or *b*) screening out shale after primary crushing, to eliminate the shale waste that could not be separated during open cut mining.

Advantages: - if successful the processes could significantly upgrade the deposit.

6. Recalculation of reserves by different methods. This would be best achieved by putting all drillhole data on computer.

Advantages: - the deposit may be upgraded by using ^a different method of grade calculation; *or downgraded*

- will not require a large capital outlay; *Expense in time (personal) computer time*

- a statistical analysis of the available data could be more easily carried out. *Doubtful if real changes were estimated significantly*

3. CONCLUSIONS/RECOMMENDATIONS:

A) Further intensive drill testing of the deposit is not recommended at this stage in view of the doubt cast on grades estimated by this method. This comment does not apply to diamond drill testing for oxide and sulphide reserves along the fault zone between the north and south blocks and the untested soil geochemical and magnetic anomalies outside of the two main blocks.

B) The proposal to drill out mine and mill a small block of ore is probably the most cost effective way of attempting to solve the drill grade versus actual grade problem. This programme could be accompanied or followed by bulk sampling of existing and newly developed underground workings.

ESTIMATED COSTS OF ALTERNATIVE PROGRAMMES

Programme	Est. Time	Est. Cost		
1. 8000m percussion drilling on 25m centres to 100 level.	6-8 months + (one rig)	Drilling (\$50/m)	\$400,000	
		Analyses (Sn, Cu, Ag, W)	40,000	
		Wages/Salaries	45,000	
		Messing/Accommodation	5,000	
		Transport	5,000	
		Other	10,000	
			<u>\$505,000</u>	✓ DV
2. Drill out, mine, mill 2000 tonne ore block (9m ³).	8-10 months +	Drilling (\$150/m +)	30,000	
		Analyses	10,000	
		Mining (open cut) \$20/T	40,000	
		Milling \$15/T	30,000	50,000
		Capital costs (plant, dams, power)	50,000	250 - 300,000
		Wages/Messing/Transport	60,000	
		Other	10,000	
	<u>\$230,000</u>	432,000 DV		

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GREAT PYRAMID, TASMANIA

D. Vance,
Townsville
8/11/1981

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SUMMARY

1. Grade
2. Recovery
3. Drilling
4. Cost Estimates
5. Exploration alternatives
6. Conclusions and Recommendations
7. Estimated costs of alternatives

SUMMARY

The most disturbing feature of the information available is the discrepancy between ore-grade as calculated by B.H.P. and that calculated by Aberfoyle.

The very high cost of drilling, the unreliability of samples from drilling and the "spotty" nature of the tin mineralisation suggest that the only adequate way of developing reserves is by driving and treating the spoil in a pilot plant.

This would also provide an accurate figure for recovery and would determine the parameters for mill design.

It may be possible to extend the drives into the fault zone to test for ore.

Costs are estimated at about M\$0.8, over M\$0.3 being capital cost. If much of the equipment were available within the Company, and allowing credit for tin concentrates recovered costs could be as low as M\$0.5. With unforeseen problems and/or low recovery and/or expansion of the programme, costs could rise to M\$1.

Recovery would be expected to be high with a simple coarse cassiterite ore. Ore-dressing tests have shown good recoveries on higher grade ore, but disturbingly consistent lower recoveries on lower grade material. This may be due to a "background" of essentially irrecoverable tin, as fine disseminated cassiterite, as tin silicate, or as ionic replacement in other minerals. Sec 1977
CRS 8-10/23

I would expect a higher grade concentrate than 45% Sn, possibly in excess of 60% Sn.

Resource Technology have grossly over-estimated the mine capital cost. I estimate M\$1.6 (vs M\$11.22), allowing for some use of contractors in periods of excess overburden removal. Total capital costs I estimate at M\$10.

Their operating cost estimate of \$13.50 per ton includes \$6 for administration, overheads and contingencies. Trimming a little fat here should reduce operating costs to \$12/ton or less. Using this figure and their graphs for tin concentrate revenue, an operating surplus of M\$5 p.a. from ore production of 0.5 M.t.p.a. would be attained at the following grade-recovery figures.

<u>Recovery (%)</u>	<u>Grade (%Sn)</u>
80	0.236
75	0.252
70	0.270
65	0.290
60	0.317

On a capital cost of M\$10, an operating surplus of M\$5 p.a. is satisfactory for a mine life of 8 years or less.

Exploration of other prospects in the area is unlikely to cheaply indicate significant reserves. I therefore consider that the Great Pyramid deposit remains the key to establishing a profitable operation.

The type of tin mineralisation, i.e. open space filling in fractures, suggests that diamond drilling might seriously undervalue the deposit by loss of grains of cassiterite from the more friable vein material. This may be part of the explanation of the lower grades calculated by B.H.P. It would be interesting to compare sludge assays with core assays.

I feel that percussion drilling (by air-track drill) would be much better for indicating open cut reserves in the Brock's and Brock's West areas, and in the Great Pyramid North Extension area. It may also have some application in the Cramp's area and for collaring of deep diamond drill holes.

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Owing to my ignorance of the deposit, and lack of detailed information, the foregoing should be regarded as tentative suggestions, rather than as definite recommendations.

1. GRADE

This is obviously a critical factor and the great difference (0.31% Sn vs 0.22% Sn) for the Aberfoyle and B.H.P. calculations suggests that there is a serious sampling problem.

The cassiterite occurs in open fracture fillings which are vuggy and friable at least in the oxidised zone. Some of this material may be lost from drill-core, which would cause under-valuation. A comparison of core assays with sludge assays should indicate the degree of this problem.

For the same reason, percussion drilling might over-value the deposit by including richer vein material from outside the nominal periphery of the hole in the sample. This is not certain as material lost from the air-stream into open fissures would be expected to be enriched in cassiterite.

I doubt if any type of drilling will provide accurate samples, and think that driving will be required to determine grade with reasonable confidence.

2. RECOVERY

This is also critical, and again the information appears somewhat unreliable. Nevertheless there is a definite tendency for lower-grade ore to show lower recoveries in tests. Recoveries with higher grade ore have been satisfactory. Possibly very fine grains of cassiterite were disseminated in the matrix of the (pre-quartzite) sandstone at an early stage in the genesis of the deposit. Alternative possibilities are that a small but reasonably constant amount of tin occurs as tin silicate or as ionic replacement in other minerals.

Sulphide - containing cassiterite ores are not necessarily much more difficult to treat than oxidised ores, depending on grain size and mineral association. They may require a

floatation cell to float sulphides off the concentrates, but this is not very expensive.

3. DRILLING

Drilling has been quite unsatisfactory. Costs have been very high, samples obtained unreliable and many holes failed to reach their targets.

As potential ore-blocks are restricted in size, they would need to be close to the surface for open-cut mining to be considered. I think therefore that the best approach would be to indicate reserves by relatively shallow (? 30m) percussion drilling using an air-track drill, and to prove any reserves so indicated, if necessary, by adits + drives.

Diamond drilling would obviously be required to indicate deeper reserves or to better define the structural or stratigraphic parameters of a deposit where knowledge of these is important. Even here, collaring the holes by percussion drilling might save considerable money.

Few tin deposits are readily drill-provable.

4. COST ESTIMATES

Resource Technology have estimated capital and operating costs as follows :

SCALE 0.5 Mtpa

<u>CAPITAL</u>		<u>OPERATING</u>	
Mine	11.22	Mining	2.50
Mill	5.68	Concentrating	5.00
Other	2.93	Admin. overheads,	63.00
		<i>contingency</i>	
TOTAL	<u>M\$19.83</u>	TOTAL	<u>M\$13.50</u>

The mine capital cost figure is certainly too high. I estimate as follows :

1 only	large air-track drill, c.w. compressor, rods, etc	250,000
1 only	4 m ³ wheel loader	400,000
1 only	dozer c.w. rippers	200,000
4 only	tandem drive trucks c.w. 3-axle tipping semis	350,000
1 only	grader	120,000
1 only	water-tanker	30,000
1 only	ANFO truck	25,000
2 only	4 x 4 utilities	25,000
	Diesel tanks	5,000
	Magazines, crib-room	10,000
	H.E. service bay	35,000
	Contingency	150,000
	TOTAL	<u>\$1,600,000</u>

This is a minimum figure, and would probably require some use of contractors in periods of excess overburden removal. Alternatively a larger loader and extra truck might be warranted, or pre-stripping of excess overburden might be feasible. If so, this is probably the least solution.

I haven't costed the mill in detail, but the figure seems reasonable. It should allow for 3-stage crushing with associated bins, feeders, conveyors and screens. Presumably recovery would be initially by jigs if sufficient coarse cassiterite were present, with the coarser fraction of the jig tailings going to a grinding mill, sands fed by classifiers to sand tables and slimes by cyclones to slime tables, with regrind of table middlings. With sulphide ore, a small floatation cell would presumably be required to clean up the

concentrates. This is a simple layout which allows for good recovery at good concentrate grade on simple ores.

Ancillaries do not appear to be very difficult. Connection to a HT transmission line is usually easy in Tasmania. Kelly Creek looks as if it would be a suitable site for a tailings dam, and the mill could probably be erected on one of the gently sloping hills north of this. Weathered Mathinna shale might be suitable for an earth wall for the tailings dam. The Scamander River flows permanently and a pump and short pipeline should provide make-up water at low cost. The towns of Scamander and St. Helen's are sufficiently close to the site to obviate the need for much housing.

Capital required is estimated at:

Mine	1.6
Mill	6.0
Other	2.4
Total	<u>M\$10.0</u>

The Resource Technology estimate for operating costs appears reasonable except for administration, overheads and contingencies, which appear excessive even for a small mine. I estimate operating costs as :

Mining	2.50
Concentrating	5.00
Admin. overheads, contingency	4.50
Total	<u>M\$12.00</u>

On these figures, at an ore production of 0.5 Mtpa, an operating surplus of M\$5 p.a. would be attained at a recovered grade of 0.19% Sn, on M\$10 capital expenditure. An 8 year mine

life on a 4 Mt reserve is quite adequate

5. EXPLORATION ALTERNATIVES

Comments on memo from R. Hine, Oct, 1981.

- (i) Pattern drilling with vertical percussion. I agree with the comments.
- (ii) Close-drill and mine a small block of ground, then mill the spoil.

Other disadvantages are :

- (a) Both percussion and diamond drilling would be required, as correction factors would probably be different for each.
- (b) The holes would have to be designed as blast-holes or the large number of holes would make blasting difficult.
- (c) Edge effects e.g. sub-grade drilling and wall and face over-break could be significant and not readily quantifiable.
- (d) Correction factors may vary with depth of hole particularly for percussion drilling.
- (e) The total amount of drilling is not great, and may not yield statistically significant results.

(iii) Driving and bulk sampling.

In addition to the advantages listed adits or drives could be extended into the fault zone to test for ore, and, if desired, continued along the fault zone to test its variability.

Adits and drives can be opened at any level desired, but obviously at extra expense.

- (iv) Diamond drill smaller potential ore zones in Great Pyramid area.

I think that small potential open cut areas would be better tested by percussion drilling, with subsequent adits + drives for proved reserves if necessary.

Mineralization along the fault would be more cheaply and better tested by extending adits or drives, possibly with long-hole sludge drilling from drive faces.

- (v) Investigate the potential for ore-sorting or differential screening.

I could not comment without seeing the ore, but doubt if much improvement of the economics is likely.

- (vi) Re-calculation of reserves.

Certainly the wide difference between the B.H.P. and Aberfoyle figures would suggest this.

It does not however, address the fundamental problem of unreliable samples. There is no mathematical method, computerised or otherwise, which will produce reliable ore reserves from unreliable samples.

6. CONCLUSIONS AND RECOMMENDATIONS

- (A) I agree that intensive drill testing is not warranted. I think that near-surface reserves would be better tested by percussion drilling, and that the fault zone would be better tested by driving + sludge hole drilling.
- (B) I would not recommend this for the reasons listed under (ii) above.
- (C) The south-west block at Great Pyramid is the larger and apparently the richer. It would presumably therefore be worked first and hence is the most important in the overall economics. I think it should be thoroughly tested by driving and milling the spoil. If this block shaped up well enough, indicated reserves from other areas might be adequate to justify an operation.

7. ESTIMATED COSTS OF ALTERNATIVES

<u>Programme</u>	<u>Estimated Time</u>	<u>Estimated cost</u>			
(i) 8000m percussion drilling	6 - 8 mos	Drilling @ \$50/m	\$400,000		
		Other	105,000		
		Total	<u>\$505,000</u>		
(ii) Drill, mine, mill 2000t ore-block	6 mos	100m D/drilling @ \$150/m	\$ 15,000		
		100m P/drilling @ \$ 30/m	3,000		
		Mining @ \$10/t (incl. stripping)	20,000		
		Milling @ \$25/t	50,000		
		Capital cost mill etc.	280,000		
		Wages, messing, transport	60,000		
		Other	10,000		
		Total	<u>\$438,000</u>		
(iii) 900m U/G drives for c, 8000t ore $\left\{ \begin{array}{l} 2.2 \times 1.5 = 3.3\text{m}^2 = 8.6\text{tm}^{-1} \\ @ \text{R.D.} = 26\text{tm}^{-3} \end{array} \right\}$	12 mos	8000t mining @ \$33/t	\$260,000		
		8000t milling @ \$25/t	200,000		
		Analyses	20,000		
		Capital cost mill etc.	280,000		
		Capital cost mine	40,000		
		Wages, messing, transport	100,000		
		Other	20,000		
		Total	<u>\$920,000</u>		
		(iv) Testing new areas and fault zone	3 mos	200m driving @ \$300/m	\$ 60,000.
				250 m sludge holes @ \$8/m	2,000
1000m percussion drilling @ \$20/m	20,000				
Analyses	10,000				
Wages, messing, transport	30,000				
Other	10,000				
Total	<u>\$132,000</u>				

Extra if D/drilling required.

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Aug. 1981

TABLE 2
PYRAMID PROSPECT
SUMMARY OF RESERVES
210 LEVEL TO 90 LEVEL

	Probable		Possible		Potential	
NORTH BLOCK (E)	1,012,672	.21	442,101	.20	1,928,964	.15
SOUTH BLOCK (W)	2,219,753	.22	429,229	.25	2,254,541	.17
TOTALS	3,232,425	.22	871,330	.22	4,183,505	.16

4,103,755

- Total calculated from drilling 4.10mT at 0.22% Sn.
- Total for deposit to 90 level 8.29mT at 0.19% Sn.
- Contained tin 15,746 tonnes.

NORTH BLOCK

Level	Probable		Possible		Potential	
	T	%Sn	T	%Sn	T	%Sn
210	9,548	.13	1,855	.10	30,608	.15
	247	.24	-	-	-	-
	2,995	.35	398	.30	-	-
200	46,907	.15	15,980	.16	51,304	.15
	15,088	.25	-	-	12,455	.20
	14,143	.35	11,395	.37	-	-
190	131,352	.15	12,720	.16	114,772	.15
	55,332	.24	-	-	8,586	.20
	24,001	.47	-	-	-	-
	1,590	2.04	-	-	-	-
180	153,444	.15	37,630	.12	132,103	.15
	43,822	.23	-	-	5,300	.20
	4,452	.34	-	-	-	-
170	54,697	.16	41,817	.13	297,993	.13
	86,515	.24	6,413	.22	48,495	.21
	30,722	.34	-	-	-	-
160	55,279	.14	77,592	.13	328,600	.12
	89,703	.25	21,200	.23	57,240	.20
	6,890	.36	-	-	-	-
150	65,086	.14	10,097	.13	233,200	.15
	15,396	.23	33,655	.23	-	-
	5,035	.33	-	-	-	-

Level	Probable		Possible		Potential	
	T	%Sn	T	%Sn	T	%Sn
140	49,891	.14	5,750	.13	133,693	.15
	37,472	.24	49,635	.23	63,865	.20
	6,387	.33				
130	4,876	.11	32,065	.14	135,150	.10
	1,802	.20	28,143	.63	41,075	.40
120	-	-	34,503	.15	121,900	.15
110	-	-	5,353	.10	49,025	.15
			15,900	.20		
100	-	-	-	-	-	-
90	-	-	15,688	.21	63,600	.15
TOTALS	571,080	.15	275,362	.13	1,691,948	.14
(North Block)	345,377	.24	126,803	.23	195,941	.20
	94,625	.38	39,936	.55	41,075	.40
	1,590	2.04				
	(1,012,672	.21)	(442,101	.20)	(1,928,964	.15)

SOUTH BLOCK

Level	Probable		Possible		Potential	
	T	%Sn	T	%Sn	T	%Sn
200	22,561	.14	11,395	.20	-	-
	7,659	.23				
	4,929	.30				
190	123,085	.13	-	-	25,149	.15
	22,083	.23				
	795	.30				
180	131,486	.13	-	-	28,832	.15
	54,396	.23				
	108,275	.45				
170	190,129	.14	9,010	.15	3,419	.15
	98,880	.25				
	43,124	.38				
160	200,790	.13	33,792	.15	56,206	.11
	72,825	.24				
	101,102	.36				
150	181,171	.14	21,624	.15	12,455	.10
	98,607	.23				
	84,121	.39				
	1,961	.97				
140	93,108	.14	24,857	.12	329,130	.14
	57,020	.23				
	44,486	.38				
	1,193	1.23				

Level	Probable		Possible		Potential	
	<u>T</u>	<u>%Sn</u>	<u>T</u>	<u>%Sn</u>	<u>T</u>	<u>%Sn</u>
130	106,486	.13	19,558	.15	245,655	.15
	50,934	.22	-	-	48,230	.25
	50,032	.41	19,001	.37	66,780	.30
120	101,955	.15	8,374	.16	232,140	.11
	34,053	.23	3,180	.20	65,415	.20
	13,585	.36	2,756	.30		
110	42,602	.14	37,524	.11	164,830	.13
	23,868	.23	16,218	.26	94,870	.20
	4,505	.38	9,275	.38		
100	12,614	.14	34,742	.14	205,905	.15
	1,670	.21	6,175	.25		
	4,770	.38				
90	28,390	.13	3,180	.18	284,080	.15
	3,657	.40	17,826	.21	15,900	.56
			2,650	2.80		
TOTALS	1,234,377	.14	192,661	.14	1,587,801	.14
(South Block)	521,995	.23	106,037	.24	498,028	.21
	463,381	.40	130,531	.43	168,712	.32
	(2.22mT	.22)	(0.43mT	.25)	(2.25mT	.17)

11th September, 1981.

MEMORANDUM TO: MANAGER EXPLORATION

 c.c. C. Palethorpe - Camberwell
 A. Clarke - Brisbane
 R. Hine - Hobart

FROM: MANAGER RESOURCE TECHNOLOGY

GREAT PYRAMID TIN PROSPECT

Please find attached a series of curves describing what we consider to be reasonable tin exploration targets at Great Pyramid. The curves describe grade-tonnage combinations at various mill recoveries, which will yield discounted cash flow rates of return in excess of 15%, assuming that the specific conditions attached hold.

As you can see, the project is quite sensitive to metallurgical recovery, and in fact further analysis of the project would be of little benefit without a better estimate of the expected mill recovery.

Within the range under consideration, you will note that the grade-tonnage relationship become more critical at reserve tonnages below 5 million, as the scale of the required operation decreases.

In addition, the before tax operating surplus of an operation based on less than 5 million tonnes at the grades and recoveries indicated is likely to be less than \$5 million per annum and is probably too small to warrant our interest.

May we suggest that should the prospect continue to be marginal or sub-marginal, it should be farmed out to an operating Tasmanian tin producer, e.g. Renison or Aberfoyle. At this stage it is unlikely that Resource Technology will be able to cope with such a demanding project, unless there is an agreement to a significant increase in our staff.

MANAGER RESOURCE TECHNOLOGY.

HMCK:ES
Melbourne
6287

ASSUMPTIONS1. Revenue:

B.H.P. long range forecast is for tin prices in the range of US\$7.50 to US\$6.50 per pound escalating at C.P.I. over the next decade (1981 money values). At the Business Plan exchange rate of US\$1.2 = A\$1, this is a range of A\$13,750 to A\$11,917 per tonne of tin. For this exercise, A\$12,800 per tonne has been used. No allowance has been made for revenue from by-products (possibly Ag, Zn).

2. Smelter Returns:

It is assumed that concentrates will be sold to A.T.S. A.T.S. make payment on the basis of the full tin content less one unit, plus or minus 0.01% for each 0.1% variation in the assay below or above 74%. It is assumed that there are no impurities which would attract penalties.

3. Metallurgical Performance:

It is assumed that a concentrate of 45% Sn will be produced. Three cases of mill recovery have been examined: 60%, 70% and 80%. Although metallurgical test work is far from complete, historical evidence would suggest recoveries in the order of 65%.

4. Operating Costs:

Cost estimates have been based on a Renison study on the nearby Anchor tin prospect, dated January 1980, except for mining which is based on Mt. Newman Mining Company's current budget. An ore to waste ratio of 1:1 is assumed.

Element	0.2 mtpa Renison 1979 Estimate	Escalated to 1981 Values	Great Pyramid Estimate		
			.5 mtpa	1 mtpa	1.5 mtpa
Mining		\$2.50/t	\$2.50/t		
Concentrating	\$4.38	\$5.39	\$5.00		
Admin. and Overheads	\$3.07	\$3.78	\$3.00		
Contingency	\$2.30	\$2.83	\$3.00		
TOTAL			\$13.50	\$12.50	\$11.50

5. Capital Costs

Costs quoted in the Renison study have been escalated for time and scale.

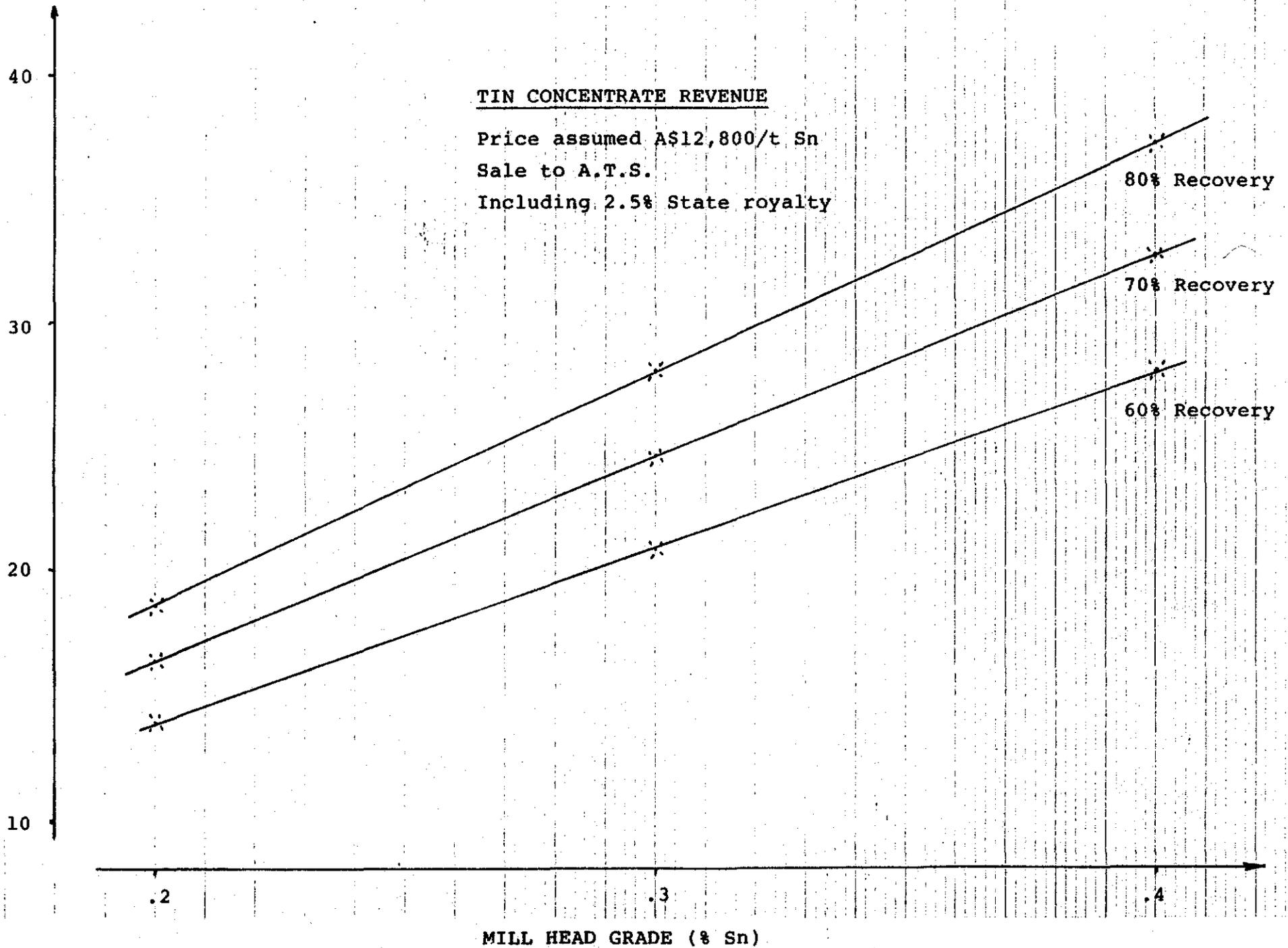
	Renison		Great Pyramid		
	<u>1979</u>	<u>1981</u>	<u>.5 mtpa</u>	<u>1 mtpa</u>	<u>1.5 mtpa</u>
Mine	5.27	6.48	11.22	17.01	21.68
Mill	2.67	3.28	5.68	8.61	10.97
Other	1.37	1.69	2.93	4.44	5.66
TOTAL	<u>9.31</u>	<u>11.45</u>	<u>19.83</u>	<u>30.06</u>	<u>38.31</u>

Mine establishment costs may have been over-estimated, as some underground mining was planned in the Renison study.

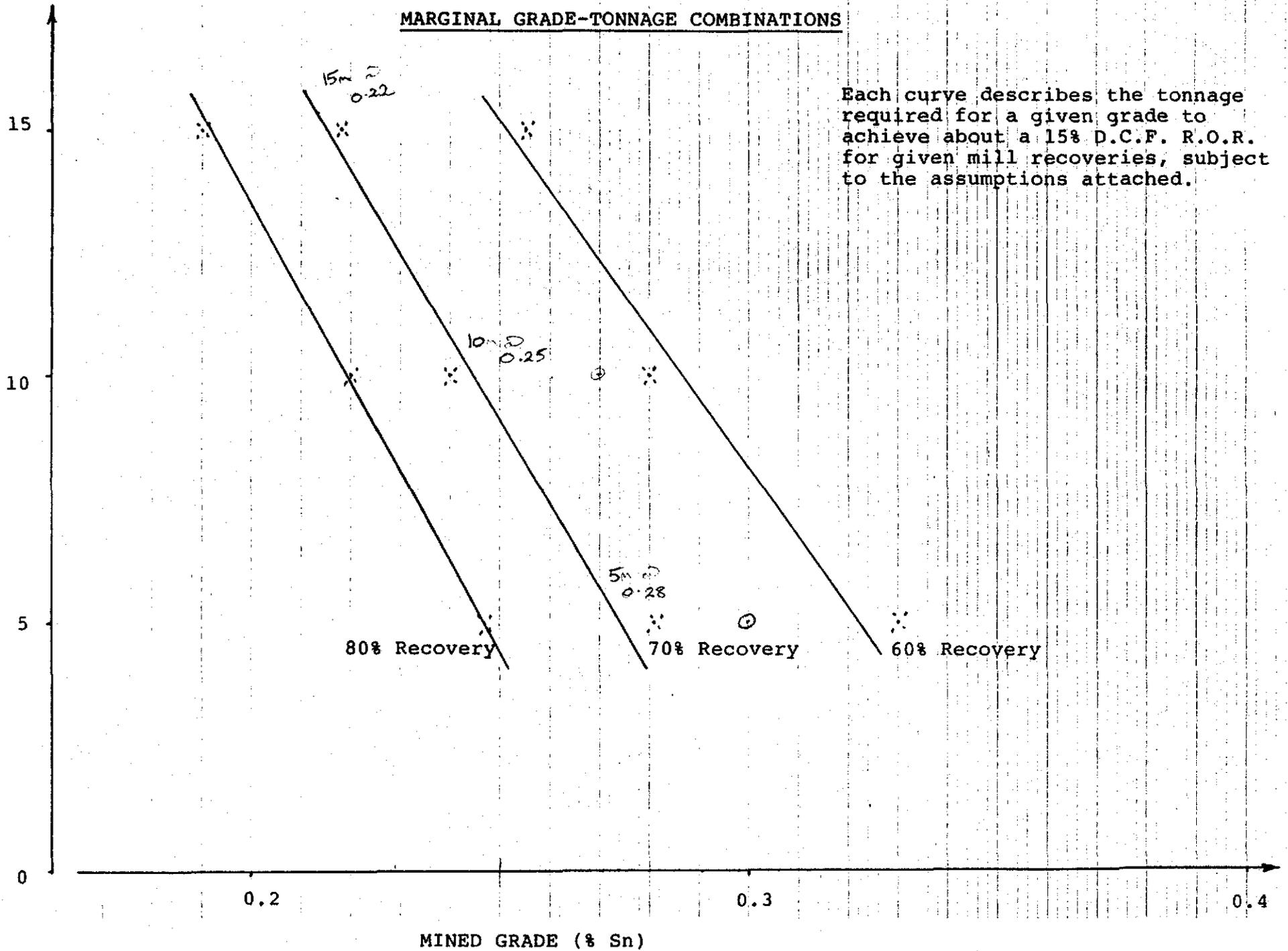
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REVENUE PER TONNE MILLED (A\$/T)



MARGINAL GRADE-TONNAGE COMBINATIONS





THE BROKEN HILL PROPRIETARY CO. LTD.

HOBART EXPLORATION OFFICE

MEMO TO : A Clarke

OUR REF : RH:RF

DATE : 17 December 1981

cc C Palethorpe

B Vivian

D Steele

H McKinnon

A Goode

YOUR REF :

FILE :

DATE :

SUBJECT : Pyramid Ore Reserve Calculation: Comparison of Methods

In order to demonstrate the effect of varying the method of calculation of ore reserves on tonnage and grade, I have re-calculated reserves for the 170 level (selected at random) by the use of rectangular areas of influence on drillholes rather than by triangles.

Comparison of results in the table below shows that for this level the use of rectangles resulted in a decrease in the calculated tonnage, above the 0.1% cut-off level, of 20% (100,000 tonnes in 500,000) and an increase in calculated grade of 20% - up to 0.25% Sn from 0.20% Sn.

Applying these corrections to the previously calculated total for the deposit (4.10mT at 0.22% Sn) gives a drill indicated reserve of 3.3mT at 0.26% Sn. There is at least a further 5mT of untested ground adjacent to the drilled mineralization.

If bulk underground sampling were to increase the grade by a further 20%, as evidence to date suggests is quite likely, a grade of around 0.30% Sn could be achieved.

SUMMARY OF RESERVES 170 LEVEL:**Method A: Triangles**

<u>North Block</u>		<u>South Block</u>		<u>Totals for Level</u>	
T	%Sn	T	%Sn	T	%Sn
96,514	.15	199,139	.14	295,653	.14
92,928	.24	98,880	.25	191,808	.25
30,722	.34	43,124	.38	73,846	.36
<u>220,164</u>	<u>.21</u>	<u>341,143</u>	<u>.20</u>	<u>561,307</u>	<u>.20</u>

DEALT WITH

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Method B: RectanglesNorth Block

T	%Sn
72,373	.14
40,095	.26
60,597	.37

173,065	.25
---------	-----

South Block

T	%Sn
128,605	.13
42,640	.23
112,043	.39

283,288	.25
---------	-----

Totals for Level

T	%Sn
200,978	.13
82,735	.24
172,640	.38

456,353	.25
---------	-----

Rick Hine



HOBART EXPLORATION OFFICE

MEMO TO : D A Clarke

OUR REF : DS:RF

DATE : 27 January 1982

cc R Hine
A Goode
H McKinnonB Vivian
C Palethorpe
D Vance

YOUR REF :

FILE :

DATE :

SUBJECT : Pyramid - Effect of sludges on grade/tonnage calculations

Serious undervaluing of mineralization at Great Pyramid may result from diamond drilling due to the liberation of cassiterite from the friable vein material. Quantitative analysis of Sn results from sludge samples collected during the 1980-81 drilling programme was undertaken to ascertain the extent of these losses and their possible effects on the grades returned.

Comparison was made of 320 core/sludge assay pairs with a lithological subdivision into shale (logged as shale and/or siltstone) and quartzite (logged as quartzite and/or sandstone, including minor shale interbeds) categories. The weightings of assays was based on the relative volume proportions of sludge to core depending on core size of the hole (ie HQ3 or NQ3). Because of statistical uncertainty (too few data points) in the comparison of sludge to core assays at grades greater than .15%Sn, recalculation of core grades was restricted to those $\leq .15\%Sn$; grades greater than .15%Sn were assumed to be unaffected by weighting of sludge assays.

Recalculation (use of sludge assays) produced the following effect on grades:

Table 1

Effect on Downhole Average

Hole	Previous Average (%Sn)	Recalculated Average (incl sludges)	% Increase in Downhole Average	Hole Length (metres)	No of Core Samples
BPD 1	.10	.13	30	101.20	51
2a	.11	.14	27	19.65	10
3	.10	.13	30	100.03	50
4	.06	.09	-	125.55	63
5	.07	.09	-	100.23	48
7	.13	.16	23	139.55	66
8	.03	.04	-	103.08	52

DEALT WITH

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70-80 only

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Hole	Previous Average (%Sn)	Recalculated Average (incl sludges)	% Increase in Downhole Average	Hole Length (metres)	No of Core Samples
BPD 9a	.13	.16	23	82.20	41
10	.15	.18	20	102.95	46
11	.06	.08	-	219.75	110
12	.02	.03	-	61.48	27

In the light of these increases the contribution of data from BPD 1-7 to the ore reserve calculation was re-evaluated. These holes in the North Block were chosen because of their locations away from other grade/tonnage influences (drillholes, adits, etc) to demonstrate the effect of the recalculation.

Results are summarized in Table 2.

Table 2

Re-evaluation of reserves - Possible Category, North Block

Source	Previous Calculation		"Sludge-weighted" Calculation	
	T	%Sn	T	%Sn
BPD 1-7	325,595	.20	(325,595 216,426)	.24 .12
Other	116,506	.20	116,506	.20
TOTAL	442,101	.20	658,527	.19

The weighted inclusion of sludge assays had two obvious effects:

- i) 20% increase in grade of tonnage used in the earlier calculation;
- ii) Additional contribution of 216,426T at .12%Sn of material previously considered below "cut-off" (.1%Sn).

The net effect was to increase the tonnage of the Possible category by 50% with a marginal decrease (5%) in grade. The decrease in grade is directly related to the inclusion of the large block of very low grade material, but is indirectly related to the method of calculation, in that a larger increase in the grade of the "previously used" material could be obtained if core grades greater than .15%Sn had been weighted.

The recalculation also proved a 2% increase in grade of material from BPD 1-7 included in the Probable Category, viz: 86,075T from .15%Sn to .19%Sn.

If increases of this magnitude were applied to grades returned from the diamond drillholes in the South Block - BPD 8-11, GPY 1-6, MD 1-4 - significant increases in the overall grade and possibly tonnage of this block could occur.

The results presented on the preceding pages clearly indicate a significant loss of cassiterite mineralization from core during drilling, resulting in an undervaluation of grades returned from holes of this type. These losses are apparently greater in the softer (shaley) lithologies. Given the numerous problems encountered during drilling and in the appraisal of drillhole results, it can only be concluded that bulk testing is required to accurately prove the grades of mineralization in this deposit.

David Steele
Geologist

27 January 1982

Appendix to Memo: Effect of sludges on grade/tonnage calculations1. Method of Calculation

320 core/sludge assay pairs were used in the calculation of average grade increases for various lithology grade intervals. The results are tabulated below.

Table 1Average grade increases for core grade/lithology

Core Grade	Lithology	
	<u>Quartzite</u>	<u>Shale</u>
0.0-0.02*	5.28 (43)	6.99 (30)
.021-0.05	2.21 (67)	2.79 (35)
.051-0.10	1.36 (53)	2.06 (12)
.101-0.15	1.52 (25)	2.52 (5)
.151-0.20	.82 (10)	.57 (5)
.201-0.25	.75 (8)	-
.251→	1.08 (14)	-

* ratios of 15 or greater omitted from calculation - would give unrealistically high increases.

(5) figures in brackets indicate the number of pairs used in determining average grade increases.

- no pairs available for shale with core grades >.20%Sn.

Increase tabulated above are averages of ratios (sludge assay ÷ core assay) for each given core grade over 2 metre intervals. Ratios show a very strong bias towards the lower grade end, trends being parallel in both lithologies, indicating that the bias is an artifact of drilling and suggests that at very low grades, cassiterite present is likely to be very fine and probably more easily liberated

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than at higher grades (coarser grain sizes?). It could also indicate that a certain amount of fine cassiterite is always liberated but decreases in significance as overall grade increases. The trend of ratios in shale being higher compared with those in quartzite over the same core grade ranges is consistent with the idea of a higher proportion of liberation of material from the softer lithologies, a trend emphasised by higher sludge grades returning from horizons of low RQD and/or core recovery.

The bias in the number of samples away from the higher grade end is mainly a consequence of the lack of water return (hence sludge samples) in areas of better mineralization, and partly due to the locations of many BPD series holes outside the defined high grade blocks.

The weighting of sludge assays to core assays was restricted to core grades below .15%Sn in view of the statistical uncertainties in the average grade increases above .15%Sn (too few data points), and partly because above this grade, sludges apparently have a less significant contribution, although this may be a consequence of having too few sample points.

Recalculated (weighted) core grades were achieved via the following formula:

$$\begin{aligned} \text{Recalculated core grade} &= (\text{original grade} \times \text{average grade increase} \times \\ (\%Sn) & \quad \quad \quad \text{volume proportion of sludge}) \\ &+ (\text{original core grade} \times \text{volume proportion} \\ & \quad \quad \quad \text{of core}) \end{aligned}$$

The average grade increase was obtained from Table 1 for each core grade. The volume proportions of sludge and core were calculated from:

$$\text{Volume of sludge} = \text{Volume of Drillhole} - \text{Volume of core}$$

$$\text{Proportion sludge} = \frac{\text{Vol of Drillhole} - \text{Vol of core}}{\text{Vol of Drillhole}}$$

$$\text{and Proportion core} = \frac{\text{Vol of core}}{\text{Vol of drillhole}}$$

$$\begin{aligned} \text{For HQ3 size: Proportion of sludge} &= 0.5961 \\ \text{core} &= 0.4039 \end{aligned}$$

$$\begin{aligned} \text{NQ3 Proportion of sludge} &= 0.6445 \\ \text{core} &= 0.3555 \end{aligned}$$

The proportioning assumes 100% sludge recovery. This was rarely achieved with recovery generally between 10 and 60%. Thus due to only partial water recovery,

it is quite likely that the sludge grades returned are minimums, and given better water recovery, sludge grades could be significantly higher.

Calculations assume an average specific gravity for rock of 2.75 tonne per m.

2. Factors affecting calculated grade/tonnage

a) Direct effects on sludge grade.

Undervaluing of the sludge sample is almost certain through a loss of mineralization either from only partial water return, resulting from material remaining down the hole or from a failure to collect it because of a poor collection set-up or the presence of slime material.

Minor effects on sludge grade could result from contamination either by the partial transfer of material from one sludge sample to the following one causing a gradual "tailing off" of grades (especially after an extremely high one), or by the incorporation of mineralization from soft, open fractures in the wall rock, most likely in the softer lithologies.

b) Effects on calculations.

The biggest factor affecting the calculated grade is the limiting of calculations to core grades less than .15%Sn on statistical grounds. This invariably leads to an increase in the amount of very low grade material above "cut off" but no increase in the grade or tonnage of better (+ .20%Sn) mineralization, with a net effect of increasing tonnage but decreasing the average grade because of the bulk of low grade material produced.

A very minor effect could be that of delayed recovery of the sludge resulting from the time lag between an interval being cored and the sludge from that interval being collected. This will tend to become more of a problem as water return decreases and as hole length increases. This problem would cause variations in the average grade increase calculated since the sludge assay used would not be from the same interval as the core grade selected.